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PRODUCTION OF LEAD AND ZINC CONCENTRATES FROM THE FARO DEPOSIT OF CURRAGH RESOURCES CORPORATION, YUKON TERRITORY

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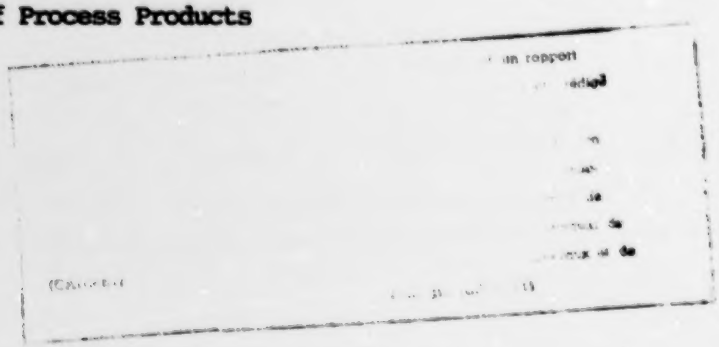

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MICROMEDIA

PRODUCTION OF LEAD AND ZINC CONCENTRATES FROM THE FARO DEPOSIT OF CURRAGH RESOURCES CORPORATION, YUKON TERRITORY

by

A.I. Stemerowicz* and G.W. Leigh**

SUMMARY

A short bench-scale investigation was carried out on the Faro ore to produce a set of suitable samples for image analysis studies. These test samples were to be representative of the optimum recoveries which could be achieved compatible with acceptable concentrate grades. After optimizing the primary grind and selective flotation test procedure, Test A-14 was carried out to provide the required samples. Results are summarized below:

	Grade %		Recovery %	
	Zn	Pb	Zn	Pb
Lead conc	5.28	54.18	4.2	83.0
Lead cl tails (1+2+3)	7.43	1.74	12.7	5.7
Zinc conc	52.83	0.78	74.8	2.1
Zinc cl tails (1+2+3)	6.42	0.83	5.7	1.4
Zinc ro tail	0.12	0.20	2.6	7.8
Feed (calc'd)	3.92	2.04	100.0	100.0

These results were obtained at a primary grind of 80% minus 47 μm. Regrinding of the lead rougher concentrate prior to cleaning was an essential step in achieving acceptable lead concentrate grade but the zinc rougher concentrate could be cleaned to a satisfactory grade without regrinding.

Keywords: Mineral processing, Concentration, Faro deposit, Lead-Zinc ore, Yukon

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PRODUCTION DE CONCENTRÉS DE PLOMB ET DE ZINC À PARTIR DE MINERAI PROVENANT DU GISEMENT DE FARO DE LA CURRAGH RESOURCES CORPORATION (TERRITOIRE DU YUKON)

par

A.I. Stemerowicz* et G.W. Leigh**

RÉSUMÉ

Une brève étude a été effectuée à l'échelle du laboratoire en vue de produire, à partir du minerai de Faro, une série d'échantillons pouvant être soumis à des analyses d'image. Ces échantillons devaient être représentatifs des taux optimaux de récupération qui pourraient être obtenus tout en étant compatibles avec des teneurs acceptables du concentré. Après avoir optimisé le premier broyage et la méthode d'essai de flottation sélective, on a effectué l'essai A-14 pour obtenir les échantillons requis. Les résultats obtenus sont résumés ci-après:

	Teneur (%)		Récupération (%)	
	Zn	Pb	Zn	Pb
Conc. de plomb	5,28	54,18	4,2	83,0
Rés. épurés de plomb (1+2+3)	7,43	1,74	12,7	5,7
Conc. de zinc	52,83	0,78	74,8	2,1
Rés épurés de zinc (1+2+3)	6,42	0,83	5,7	1,4
Rés dégrossissage de zinc	0,12	0,20	2,6	7,8
Charge (calculée)	3,92	2,04	100,0	100,0

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Ces résultats ont été obtenus avec un broyage primaire donnant un produit dont 80 % possédait une granulométrie de -47 µm. Il s'est révélé essentiel de rebroyer le concentré de dégrossissage de plomb avant l'épuration, pour obtenir une teneur acceptable pour le concentré de plomb, mais on a pu épurer le concentré de dégrossissage de zinc de manière à obtenir une teneur acceptable sans rebroyage.

Mots clés: traitement des minerais, concntration, gisement de Faro, minerai de plomb-zinc, Yukon

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INTRODUCTION

The object of the bench-scale concentration tests was to produce a set of test samples for quantitative mineral liberation studies by image analysis. These test samples were to be representative of the optimum lead and zinc recoveries which could be achieved accompanied by acceptable grades of lead and zinc concentrates. For this purpose a 100-kg sample of ore from the Faro deposit was received from the Process Mineralogy Section on January 7, 1987. This ore sample, which was in the form of minus 16 mm rod mill feed, was supplied by the Curragh Resources Corporation and was taken on October 2, 1986. At the time the investigation was begun the only useful information available to the authors regarding the mineralogy of the ore was that it contained some marcasite and was therefore susceptible to rapid oxidation.

OUTLINE OF INVESTIGATION

Crushing and Sampling

The minus 16 mm rod mill feed was first riffled into quarters. Three of the quarters were stored for future use while the remaining quarter was crushed in stages to minus 10 mesh (1680 μm). The minus 10 mesh material was then divided into 20 equal portions using a carousel-type divider. One of these portions was chosen at random as a head sample while the weights of the remaining portions were adjusted to provide 2000-gram test charges. To prevent oxidation the test charges were stored under vacuum in a desiccator. Analysis of the head sample is given in Table 1.

Table 1. Head sample analysis

Zn	4.00%
Pb	2.08%
Cu	0.13%
Ag	35.0 g/t
Total Fe	23.40%
S	20.37%
Insol.	57.27%

Grinding

The minus 10 mesh 2000-gram test charges were ground at 65% solids either in a 7 x 14 Denver laboratory rod mill or in 2 stages in a 250 mm diam ceramic ball mill using large ceramic balls in the first stage and small ceramic balls in the second stage. To eliminate the small amount of plus 14 mesh tramp oversize produced in the first grinding stage as a result of inadequate impact by the low density ceramic balls it was necessary to replace some of the ceramic balls with heavier steel balls. In every case, even when depressants were added to the mill, the first grinding solution was discarded and fresh make-up water was added to the second grinding stage. Primary grinding data is given in Table 2.

Table 2. Primary grinding data

Test No.	Stage	Grind time min	Unit	Grinding media		K_{80}* obtained, μm	
				Wt kg	Description	Flot'n feed	Flot'n tail
A-1 to A-7	1	60	7 x 14 RM	20	Steel rods, 13 to 38 mm diam	48	-
A-8	1 2	60 30	250 mm ceramic BM as above	6.8 6.12	25-28 mm ceramic balls -13 mm ceramic balls	-	73
A-9, A-10	1 2	90 30	250 mm ceramic BM as above	6.8 6.12	25-28 mm ceramic balls -13 mm ceramic balls	-	48
A-11 to A-14	1 2	90 30	250 mm ceramic BM as above	4.61 2.19 6.12	25-28 mm ceramic balls 25 mm steel balls -13 mm ceramic pebbles	47	45
A-15	1	90	7 x 14 RM	20	Steel rods, 13 to 38 mm diam	37.5	-

*80% passing size

Flotation

Conventional selective flotation techniques were employed for the production of lead and zinc concentrates from the ore. Standard procedure was as follows: soda ash and cyanide were added to the grind, the former for alkalinity control and the latter to depress pyrite and sphalerite. The ground pulp was subjected to aerative conditioning and a lead rougher concentrate was floated off using a combination of Aerofloat 242 and isopropyl xanthate as collectors. The lead rougher concentrate was cleaned 2 or 3 times. In some tests the lead rougher concentrate was reground in a ceramic mill prior to cleaning. Copper sulphate and lime were added to the lead rougher tailing to activate the sphalerite which was floated off into a zinc rougher concentrate using sodium aerofloat as the main collector augmented by small amounts of isopropyl xanthate. To ensure optimum selectivity 1000 g/t copper sulphate was added for sphalerite activation along with lime to give a pH of about 11. The zinc rougher concentrate was cleaned 2 or 3 times without prior regrinding.

Deviations from standard procedure were as follows:

(1) $ZnSO_4$ as sphalerite depressant along with cyanide (Test A-2).
(2) Pennfloat 3 as lead collector in place of Aerofloat 242 (Tests A-4, 5 and 6).
(3) Conditioning without air prior to lead flotation (Test A-12).
(4) NaOH + H_2O_2 (Test A-8) or NaOH (Test A-10) added to first stage grinding. This practice was found to be beneficial on a massive sulphide ore containing marcasite previously investigated by the authors.

The amount of cyanide added to the grind was tried at levels of 50, 100, 250 and 500 g/t.

DISCUSSION AND EVALUATION OF RESULTS

Best Results

Test procedure and results for all tests are summarized in Table 6 for lead flotation and Table 7 for zinc flotation which are appended. The

data in these tables has been plotted to produce the concentrate grade-recovery curves shown in Fig. 1 and 2. As is graphically illustrated the best combination of lead and zinc results was obtained in Tests 11, 13 and 14. Except for small differences in the amount of collector employed, the test procedure for all these tests was identical. The results for Test 14, which was specifically carried out to provide a set of test samples for image analysis studies, are given in Table 3 followed by the reagents and conditions employed in Table 4.

Table 3. Best results (Test A-14)

Product	Wt %	Analysis %		Distribution %		Sep* eff %
		Zn	Pb	Zn	Pb	
Lead conc	3.12	5.28	54.18	4.20	82.99	78.79
Lead cl tail 3	0.30	8.24	7.51	0.63	1.10	
Lead cl tail 2	0.45	8.28	3.46	0.95	0.75	
Lead cl tail 1	5.92	7.32	1.32	11.06	3.83	
Zinc conc	5.55	52.83	0.78	74.83	2.13	
Zinc cl tail 3	0.39	28.08	1.59	2.80	0.30	
Zinc cl tail 2	0.35	7.68	1.12	0.69	0.19	
Zinc cl tail 1	2.76	3.20	0.69	2.25	0.93	
Zinc ro tail	81.16	0.12	0.20	2.59	7.78	
Feed (calc'd)	100.00	3.92	2.04	100.00	100.00	
Lead ro conc	9.79	6.74	18.45	16.84	88.67	71.83
Lead ro tail	90.21	3.61	0.26	83.16	11.33	
Zinc ro conc	9.05	34.88	0.80	80.57	3.55	

*Separation efficiency = lead recovery-zinc recovery

Table 4. Reagents and conditions for Test A-14

Operation	Time min	pH	Unit used	Reagents, g/t							
				Na_2CO_3	NaCN	AF242	R343	DF250	$CuSO_4$	Lime	NaAF
Grinding - 1	90		250 mm diam BM	-	-	-	-	-	-	-	-
Grinding - 2	30		250 mm diam BM	1000	50	-	-	-	-	-	-
Aeration	20		100 mm lab aerator	-	-	25	10	-	-	-	-
Lead ro - 1	½	8.5	1000-g cell	-	-	-	-	9	-	-	-
Lead ro - 2	1½			-	-	5	5	9	-	-	-
Zinc conditioning	10	11.35	1000-g cell	-	-	-	-	-	1000	1200	-
Zinc ro - 1	½			-	-	-	10	-	-	-	39
Zinc ro - 2	½			-	-	-	-	9	-	-	-
Zinc ro - 3	2			-	-	-	-	9	-	-	13
Lead regrind*	20		250 mm diam BM	250	100	-	10	-	-	-	-
Lead cleaner - 1	2	9.6	1000-g cell	-	-	-	-	-	-	-	-
Lead cleaner - 2	1½		250-g cell	-	-	-	-	-	-	-	-
Lead cleaner - 3	1		250-g cell	-	-	-	-	-	-	-	-
Zinc cleaner - 1	1½	10.1	500-g cell	-	-	-	-	-	-	-	-
Zinc cleaner - 2	1½		250-g cell	-	-	-	-	9	-	-	-
Zinc cleaner - 3	1		250-g cell	-	-	-	-	-	-	-	-

*Lead rougher concentrate filtered and repulped in mill with fresh water to 65% solids - same ball charge used as in second stage primary grind

Abbreviations:

AF242 - Aerofloat 242
R343 - Sodium isopropylxanthate
DF250 - Dowfroth 250
NaAF - Sodium aerofloat

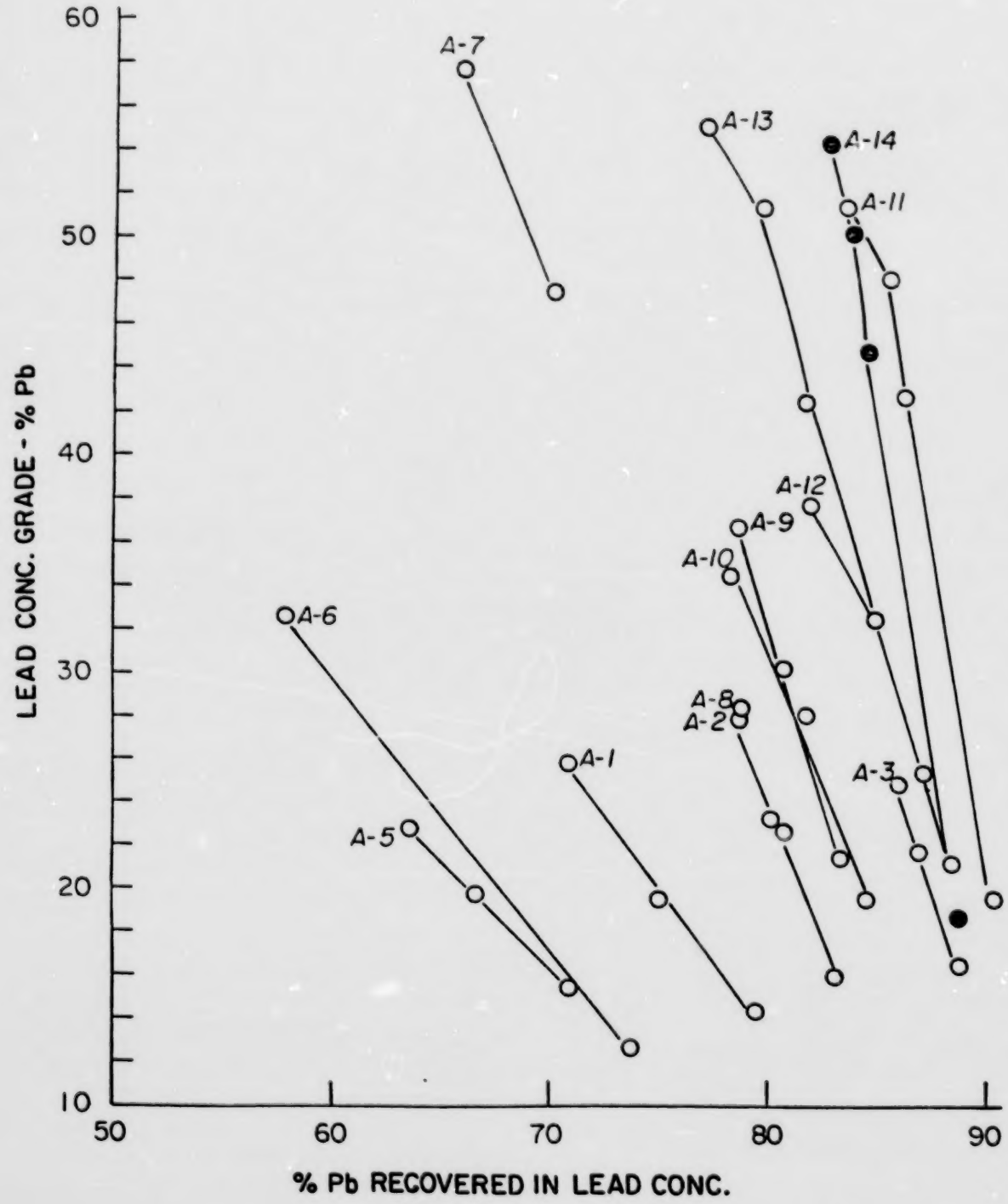

Fig. 1. Grade-recovery curve for lead flotation

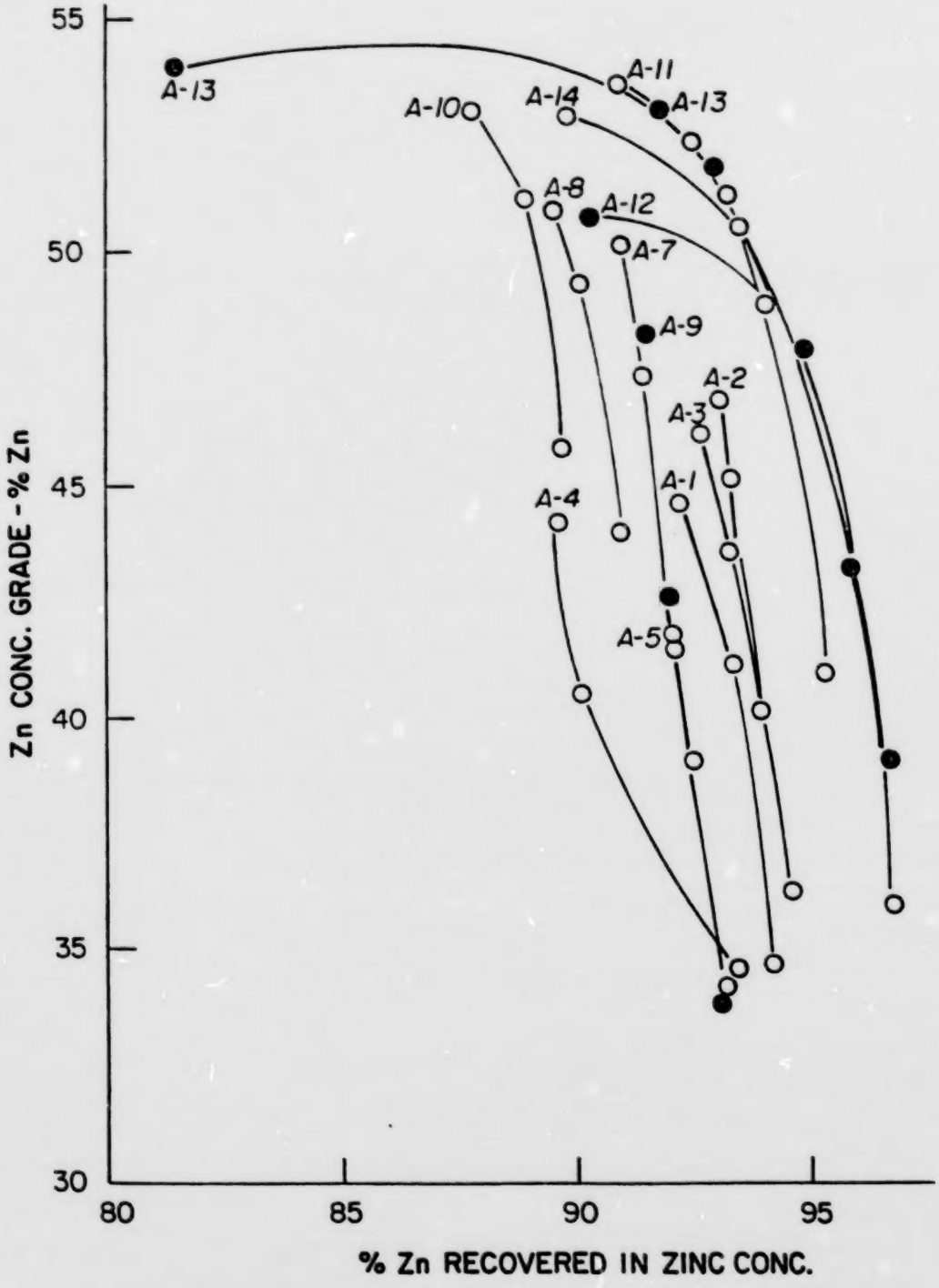

Fig. 2. Grade-recovery curve for zinc flotation

Effect of Lead Rougher Concentrate Regrind

Regrinding of lead rougher concentrate prior to cleaning was an essential step in achieving acceptable lead concentrate grade. Without regrinding, a lead concentrate grade of only 37.6% Pb was obtained in Test A-12 after 2 stages of cleaning as compared to the 50.1% grade achieved after regrinding in Test A-14. It should be pointed out that other than liberating minerals from locked particles, regrinding has a beneficial "polishing" action on minerals which removes collector coatings thus making them more susceptible to the depressing action of the cyanide added to the regrinding step.

Optimum Primary Grind

The best results were achieved at a grind of 80% minus 47 μm which was obtained by using a two-stage grind in a 250 mm diameter ceramic ball mill with a mixed charge of ceramic and steel balls in the first stage (90 min) and a charge of smaller ceramic balls as grinding media in the second stage (30 min). To determine the adequacy of this grind the zinc rougher tailing from Test A-11 was sized and the size fractions analyzed for lead and zinc. The distribution of metal losses by size fractions is given in Table 5.

The low lead and zinc losses sustained in the three coarsest size fractions of the tailing are an indication that both galena and sphalerite had been liberated to the extent that optimum rougher flotation recoveries could be achieved. Therefore, it can be concluded that a grind of 80% minus 47 μm can be considered to be optimum. Any further liberation of the valuable minerals from middling particles would most efficiently be achieved by regrinding rougher concentrates prior to cleaning. Note that the finer grind of 80% minus 37.5 μm employed in Test 15B did not result in any appreciable difference in lead rougher flotation results (compare with Tests A-11 to A-14 in Table 6).

Table 5. Metal losses in zinc rougher tailing by size fractions (Test A-11)

Size fraction	Diam μm	Wt % ret	Analysis %		Distribution %	
			Zn	Pb	Zn	Pb
+200 mesh	74	4.80	0.18	0.15	6.9	4.2
+270 mesh	53	9.20	0.16	0.14	11.7	7.6
+325 mesh	45	6.80	0.14	0.12	7.6	4.8
+400 mesh	38	20.00	0.11	0.11	17.5	12.9
CS* cone 1	26	16.22	0.09	0.15	11.6	14.3
CS* cone 2	20	12.67	0.04	0.09	4.0	6.7
CS* cone 3	14	7.22	0.04	0.10	2.3	4.3
CS* cone 4	10	5.92	0.20	0.10	9.4	3.5
CS* cone 5	8	3.67	0.07	0.13	2.1	2.8
CS O'flow	-8	13.50	0.25	0.49	26.9	38.9
Feed (calc'd)		100.00	0.13	0.17	100.00	100.00
Feed (assay)			0.12	0.16		

*Warman Cyclosizer

Effect of Other Test Variables

Cyanide addition level

The best results were obtained at a cyanide addition level of only 50 g/t. Increasing cyanide addition to the very high level of 500 g/t (Tests A-8, 9 and 10) did not have any significant effect on galena-pyrite or galena-sphalerite selectivity.

$ZnSO_4$ as sphalerite depressant

In the one test in which $ZnSO_4$ was tried in conjunction with cyanide as a sphalerite depressant (A-2) it did not have any appreciable effect on galena-sphalerite selectivity.

Dodecyl mercaptan (Pennfloat 3) as lead collector

This collector used in place of Aerofloat 242 in Tests A-4, 5 and 6 gave inferior results as measured by the much lower lead-zinc separation efficiencies obtained.

Lead conditioning without air

This was tried in Test A-12 and did not result in initially slow lead rougher flotation as is the case for other complex lead-zinc ores, especially those from New Brunswick. With these ores it is necessary to aerate the pulp to obtain heavily-mineralized, high grade froth at the start of lead rougher flotation.

NaOH and H_2O_2 additions to the first grinding stage

These reagents, which were specifically added to counteract the possible harmful effect of marcasite oxidation, had no detectable effect on results (Tests A-8 and A-10).

Grinding media

The type of grinding media employed may have a pronounced effect on sulphide flotation. On this ore there did not appear to be any differences in flotation response between grinding with steel in a rod mill and grinding with ceramic balls in a ceramic ball mill. The lead rougher flotation

results obtained using single stage rod mill grinding in Test 15B are just as good as those obtained in Test A-14 in which two-stage grinding with ceramic pebbles was employed. In some of the initial tests (A-1 and A-2) lower separation efficiencies were obtained after grinding in a rod mill but this can be attributed to the fact that collector additions had not yet been optimized.

CONCLUSIONS

The ore did not prove as difficult to concentrate as was anticipated by the authors. By carrying out only 14 flotation tests it was possible to achieve what are considered to be optimum lead and zinc recoveries which were obtained at a primary grind of 80% minus 47 μm (Test A-14). Although galena-pyrite selectivity during lead rougher flotation was poor it was found that by regrinding the lead rougher concentrate with 100 g/t cyanide, pyrite was rejected during the subsequent cleaning operation to the extent that the lead rougher concentrate was upgraded from 18.5% to an acceptable 54.2% after 3 stages of cleaning. Lead recovery in the lead cleaner concentrate was high (-83%) based on lead in the feed or 93.6% of the lead contained in the lead rougher concentrate. The rapid upgrading of the lead rougher concentrate can be attributed not only to additional liberation of galena from middling particles but also to the beneficial "polishing" action in the regrind mill. It should be possible by further cleaning to increase lead concentrate grade to 60+% Pb. In this respect a second regrind with cyanide addition (50 g/t) of the second or third lead cleaner concentrate should be considered as a means of enhancing galena-pyrite selectivity.

The zinc rougher concentrate was upgraded to a satisfactory grade of 52.8% Zn without regrinding. Zinc recovery in the zinc concentrate was only 74.8% based on the zinc in the test feed but amounted to 92.9% of the zinc contained in the zinc rougher concentrate. The difference in the two recovery figures is due in large measure to the high amount of zinc tied up in the first lead cleaner tailing (11.1%) which is not subjected to zinc flotation. In continuous plant operation it might be advantageous to divert the high-zinc first lead cleaner tailing to the zinc circuit rather than recycling it to lead rougher flotation.

The average zinc content in the sphalerite has been determined to be 62%. Therefore, the sphalerite content of the Test A-14 zinc concentrate is 85.2%. It should be possible to produce a zinc concentrate containing at least 90% sphalerite which is equivalent to about 56% zinc. It is believed that the most effective way to attain the high zinc concentrate grade would be to regrind the zinc rougher concentrate with lime prior to cleaning to take advantage of the beneficial "polishing" action.

Table 6. Summary of test procedure and results for lead flotation

Test No. date	Grind			Collector additions, g/t				Pb ro skim time	Pb ro pH	Pb ro conc RG	Lead flotation results						Sep eff %
	Type	Reagent added, g/t	K_{80} m	Cell	To condit	To Pb ro	Total				Product	Wt %	Analysis % Zn	Analysis % Pb	Distribution % Zn	Distribution % Pb	
A-1 Feb. 6	1 stage RM-60	SA 1000 NaCN 50	48	AF242 R343	25 10	20 10	45 20 65	1½	7.7	No	Cl conc 2	5.81	11.06	25.65	15.2	70.9	55.7
											Cl conc 1	8.15	9.34	19.37	18.0	75.1	57.1
											Ro conc	11.73	8.03	14.24	22.2	79.5	57.3
											Ro tail	88.27	3.74	0.49	77.8	20.5	
											Feed (calc'd)	100.00	4.24	2.10	100.0	100.0	
A-2 Feb. 10	1 stage RM-60	SA 1500 NaCN 100	48	AF242 R343	25 10	20 10	45 20 65	1½	8.0	No	Cl conc 2	6.05	7.66	27.66	11.7	78.7	67.0
											Cl conc 1	7.38	7.04	23.09	13.2	80.2	67.0
											Ro conc	11.19	6.08	15.78	17.3	83.1	57.3
											Ro tail	88.81	3.67	0.41	82.7	16.9	
											Feed (calc'd)	100.00	3.94	2.13	100.0	100.0	
A-3 Feb. 13	1 stage RM-60	SA 1250 NaCN 250	48	AF242 R343	25 10	20 10	45 20 65	1½	8.2	No	Cl conc 2	7.35	7.28	24.65	13.8	80.1	72.3
											Cl conc 1	8.49	6.86	21.53	15.0	86.9	71.9
											Ro conc	11.45	6.24	16.31	18.4	88.8	70.4
											Ro tail	88.55	3.57	0.27	81.6	11.2	
											Feed (calc'd)	100.00	3.88	2.10	100.0	100.0	

Table 6 (Cont'd)

Test No. date	Grind			Collector additions, g/t				Pb ro skim time	Pb ro pH	Pb ro conc RG	Lead flotation results						Sep eff %
	Type	Reagent added, g/t	K_{80} m	Cell	To condit	To Pb ro	Total				Product	Wt %	Analysis %		Distribution %		
													Zn	Pb	Zn	Pb	
A-4 Feb. 1	1 stage RM-60	SA 1000 NaCN 50	48	PF3 R343	25 10	17 10	42 20 62	1¼	7.8	No	Cl conc 2	20.23	3.28	6.75	17.2	66.8	49.6
											Cl conc 1	27.02	3.52	5.92	24.7	78.3	53.6
											Ro conc	34.91	3.86	5.00	35.0	85.4	50.4
											Ro tail	65.09	3.84	0.46	65.0	14.6	
											Feed (calc'd)	100.00	3.85	2.04	100.0	100.0	
A-5 Feb. 1	1 stage RM-60	SA 1000 NaCN 50	48	PF3 R343	17 10	17 -	34 10 44	2	7.9	No	Cl conc 2	5.82	8.54	22.69	11.0	63.7	52.7
											Cl conc 1	7.05	8.39	19.62	13.1	66.7	53.6
											Ro conc	9.58	14.30	15.34	30.3	70.9	40.6
											Ro tail	90.42	3.48	0.67	69.7	29.1	
											Feed (calc'd)	100.00	4.52	2.07	100.0	100.0	
A-6 Feb. 1	1 stage RM-60	SA 1000 NaCN 50	48	PF3 R343	17 10	17 -	34 10 44	2	7.8	Yes	Cl conc 1	3.52	5.48	32.55	5.3	58.0	52.7
											Ro conc	11.67	6.89	12.48	21.9	73.7	51.7
											Ro tail	88.33	3.24	0.59	78.1	26.3	
											Feed (calc'd)	100.00	3.67	1.98	100.0	100.0	

Table 6 (Cont'd)

Test No. date	Grind			Collector additions, g/t				Pb ro skim time	Pb ro pH	Pb ro conc RG	Lead flotation results						Sep eff %
	Type	Reagent added, g/t	K_{80} m	Cell	To condit	To Pb ro	Total				Product	Wt %	Analysis %		Distribution %		
													Zn	Pb	Zn	Pb	
A-7 Feb. 23	1 stage RM-60	SA 1000 NaCN 50	48	AF242 R343	25 10	20 10	45 20 65	1½	7.6	Yes	Cl conc 2 Cl conc 1 Ro tail	2.23 2.88 97.12	4.72 5.28 3.87	57.61 47.38 0.59	2.7 3.9 96.1	66.2 70.3 29.7	63.5 66.4
											Feed (calc'd)	100.00	3.91	1.94	100.0	100.0	
A-8 Feb. 25	2 stage PM-60 PM-30	NaOH 2000 H_2O_2 7500 SA 2000 NaCN 500	73	AF242 R343	25 10	- -	25 10 35	1	10.2	No	Cl conc 2 Cl conc 1 Ro conc Ro tail	5.52 7.10 10.56 89.44	6.96 6.57 6.07 3.64	28.26 22.50 15.53 0.38	9.8 12.2 16.7 83.3	78.9 80.8 83.0 17.0	69.1 68.8 66.3
											Feed (calc'd)	100.00	3.91	1.98	100.0	100.0	
A-9 Mar. 2	2 stage PM-90 PM-30	None SA 2000 NaCN 500		AF242 R343	25 10	- -	25 10 35	1	9.1	No	Cl conc 2 Cl conc 1 Ro conc Ro tail	4.53 5.64 8.24 91.76	7.92 7.71 7.22 3.78	36.55 30.10 21.29 0.38	8.8 10.7 14.7 85.3	78.8 80.8 83.4 16.6	69.9 70.1 68.7
											Feed (calc'd)	100.00	4.06	2.10	100.0	100.0	

Table 6 (Cont'd)

Test No. date	Grind			Collector additions, g/t				Pb ro skim time	Pb ro pH	Pb ro conc RG	Lead flotation results						Sep eff %
	Type	Reagent added, g/t	K_{80} m	Cell	To condit	To Pb ro	Total				Product	Wt %	Analysis %		Distribution %		
													Zn	Pb	Zn	Pb	
A-10 Mar. 4	2 stage PM-90 PM-30	NaOH 2000 SA 2000 NaCN 500		AF242 R343	25 10	- -	25 10 35	1	10.25	No	Cl conc 2	4.60	7.12	34.27	8.4	78.3	69.9
											Cl conc 1	5.90	6.87	27.90	10.4	81.8	71.4
											Ro conc	8.81	6.28	19.30	14.2	84.5	70.3
											Ro tail	91.19	3.68	0.34	85.8	15.5	
											Feed (calc'd)	100.00	3.91	2.01	100.0	100.0	
A-11 Mar. 11	2 stage PM-90 PM-30	None SA 1000 NaCN 50		AF242 R343	25 10	- -	25 10 35	2	8.3	Yes	Cl conc 3	3.24	5.64	51.39	4.8	83.8	79.0
											Cl conc 2	3.56	5.91	47.87	5.7	85.8	80.1
											Cl conc 1	4.03	6.13	42.56	6.6	86.4	79.8
											Ro conc	9.29	7.38	19.32	18.1	90.4	72.3
											Ro tail	90.71	3.41	0.21	81.9	9.6	
											Feed (calc'd)	100.00	3.78	1.99	100.0	100.0	
A-12 Mar. 18	2 stage PM-90 PM-30	None SA 1000 NaCN 50		AF242 R343	25 10	10 -	35 10 45	2	7.7	No	Cl conc 2	4.32	7.90	37.56	9.0	82.10	73.1
											Cl conc 1	5.22	8.14	32.20	11.3	85.1	73.8
											Ro conc	6.86	7.83	25.13	14.3	87.2	72.9
											Ro tail	93.14	3.47	0.27	85.7	12.8	
											Feed (calc'd)	100.00	3.77	1.98	100.0	100.0	

Table 6 (Cont'd)

Test No. date	Grind			Collector additions, g/t				Pb ro skim time	Pb ro pH	Pb ro conc RG	Lead flotation results						Sep eff %
	Type	Reagent added, g/t	K_{80} m	Cell	To condit	To Pb ro	Total				Product	Wt %	Analysis %		Distribution %		
													Zn	Pb	Zn	Pb	
A-13 Apr. 7	2 stage PM-90 PM-30	None SA 1000 NaCN 50		AF242 R343	25 10	- -	25 10 35	2	8.2	Yes	Cl conc 3	2.84	4.84	54.95	3.5	77.4	73.9
											Cl conc 2	3.14	5.09	51.36	4.1	80.0	75.9
											Cl conc 1	3.91	5.59	42.24	5.6	81.9	76.3
											Ro conc	8.54	6.51	20.88	14.2	88.5	74.3
											Ro tail	91.46	3.67	0.25	85.8	11.5	
											Feed (calc'd)	100.00	3.91	2.02	100.0	100.0	
A-14 Apr. 9	2 stage PM-90 PM-30	None SA 1000 NaCN 50		AF242 R343	25 10	5 5	30 15 45	2	8.5	Yes	Cl conc 3	3.12	5.28	54.18	4.2	83.0	78.8
											Cl conc 2	3.42	5.54	50.08	4.8	84.1	79.3
											Cl conc 1	3.87	5.86	44.66	5.8	84.8	79.0
											Ro conc	9.79	6.74	18.45	16.8	88.7	71.9
											Ro tail	90.21	3.61	0.26	83.2	11.3	
											Feed (calc'd)	100.00	3.92	2.04	100.0	100.0	

Table 6 (Cont'd)

Test No. date	Grind			Collector additions, g/t				Pb ro skim time	Pb ro pH	Pb ro conc RG	Lead flotation results						Sep eff %
	Type	Reagent added, g/t	K_{80} m	Cell	To condit	To Pb ro	Total				Product	Wt %	Analysis %		Distribution %		
													Zn	Pb	Zn	Pb	
A-15A Apr. 23	1 stage 75-RM	SA 1000 NaCN 100		AF242 R343	25 10	20 20	45 30 75	1½	8.2	No	Ro conc	16.95	3.40	8.79	15.2	75.9	60.7
											Ro tail	83.05	3.88	0.57	84.8	24.1	
											Feed (calc'd)	100.00	3.79	1.96	100.0	100.0	
A-15B Apr. 23	1 stage 75-RM	SA 1000 NaCN 100		AF242 R343	25 10	10 20	35 30 65	1½	8.35	No	Ro conc	8.68	6.44	20.38	14.0	86.8	72.8
											Ro tail	91.32	3.76	0.30	86.0	13.2	
											Feed (calc'd)	100.00	3.99	2.04	100.0	100.0	

Table 7. Summary of test procedure and results for zinc flotation

Test No.	Grind K_{80}	$CuSO_4$ addition g/t	Collector added to rougher g/t		Zn ro skim time, min	Zn ro pH	Zinc flotation results					
			R343	NaAF			Product	Wt %	Analysis %		Distribution %	
									Zn	Pb	Zn	Pb
A-1	1 stage 48	1000	-	65	1½	11.1	Cl conc 2	7.74	44.56	1.44	92.3	22.8
							Cl conc 1	8.50	41.06	1.69	93.4	29.3
							Ro conc	10.18	34.57	1.63	94.2	33.9
							Ro tail	89.92	0.24	0.36	5.8	66.1
							Feed (calc'd)	100.00	3.74	0.49	100.0	100.0
A-2	1 stage 48	1000	-	65	1½	11.2	Cl conc 2	7.31	46.75	1.08	93.2	19.5
							Cl conc 1	7.61	45.03	1.10	93.5	20.6
							Ro conc	8.61	40.04	1.10	94.0	23.3
							Ro tail	91.39	0.24	0.34	6.0	76.7
							Feed (calc'd)	100.00	3.67	0.41	100.0	100.0
A-3	1 stage 48	1000	-	65	1½	11.1	Cl conc 2	7.19	46.09	0.90	92.8	24.3
							Cl conc 1	7.67	43.50	0.92	93.4	26.6
							Ro conc	9.35	36.17	0.91	94.7	31.8
							Ro tail	90.65	0.21	0.20	5.3	68.2
							Feed (calc'd)	100.00	3.57	0.27	100.0	100.0
A-4	1 stage 48	1000	-	65	1½	11.3	Cl conc 2	7.82	44.12	1.68	89.8	28.6
							Cl conc 1	8.57	40.46	1.69	90.2	31.5
							Ro conc	10.41	34.51	1.57	93.5	35.6
							Ro tail	89.59	0.28	0.33	6.5	64.4
							Feed (calc'd)	100.00	3.84	0.46	100.0	100.0

Table 7 (Cont'd)

Test No.	Grind K_{80}	$CuSO_4$ addition g/t	Collector added to rougher g/t		Zn ro skim time, min	Zn ro pH	Zinc flotation results				
			R343	NaAF			Product	Wt %	Analysis %		Distribution %
									Zn	Pb	Zn	Pb
A-5	1 stage 48	1000	10	65	1½	11.2	Cl conc 2	7.70	41.72	3.64	92.2	42.0
							Cl conc 1	8.27	39.02	3.56	92.6	44.1
							Ro conc	9.53	34.10	3.31	93.3	47.2
							Ro tail	90.47	0.26	0.39	6.7	52.8
							Feed (calc'd)	100.00	3.48	0.67	100.0	100.0
A-7	1 stage 48	1000	10	52	2	11.4	Cl conc 2	7.03	50.08	1.51	91.1	17.9
							Cl conc 1	7.48	47.33	1.69	91.6	21.3
							Ro conc	8.61	41.39	1.90	92.2	27.6
							Ro tail	91.39	0.33	0.47	7.8	72.4
							Feed (calc'd)	100.00	3.87	0.59	100.0	100.0
A-8	2 stage 73	1500	-	39	3	11.2	Cl conc 2	6.42	50.87	1.16	89.7	19.8
							Cl conc 1	6.67	49.27	1.19	90.2	21.1
							Ro conc	7.55	43.94	1.19	91.1	23.9
							Ro tail	92.45	0.35	0.31	8.9	76.1
							Feed (calc'd)	100.00	3.64	0.38	100.0	100.0
A-9	2 stage	1500	-	39	3	11.2	Cl conc 2	7.18	48.17	1.54	91.6	29.1
							Cl conc 1	8.18	42.51	1.44	92.1	30.9
							Ro conc	10.43	33.70	1.24	93.1	34.0
							Ro tail	89.57	0.29	0.28	6.9	66.0
							Feed (calc'd)	100.00	3.78	0.38	100.0	100.0

Table 7 (Cont'd)

Test No.	Grind K_{80}	$CuSO_4$ addition g/t	Collector added to rougher g/t		Zn ro skim time, min	Zn ro pH	Zinc flotation results				
			R343	NaAF			Product	Wt %	Analysis %		Distribution %
									Zn	Pb	Zn	Pb
A-10	2 stage	1500	-	39	1½	11.2	Cl conc 2	6.10	52.99	1.12	87.9	19.9
							Cl conc 1	6.41	51.13	1.20	89.1	22.5
							Ro conc	7.21	45.79	1.19	89.8	25.0
							Scav conc	1.47	9.48	1.35	3.8	5.8
							Scav tail	91.32	0.26	0.26	6.4	69.2
							Feed (calc'd)	100.00	3.68	0.34	100.0	100.0
A-11	2 stage	1000	-	52	3	11.0	Cl conc 3	5.81	53.55	0.57	91.1	15.4
							Cl conc 2	6.05	52.31	0.60	92.7	16.8
							Cl conc 1	6.34	50.43	0.62	93.6	18.3
							Ro conc	7.97	40.88	0.62	95.4	23.0
							Ro tail	92.03	0.17	0.18	4.6	77.0
							Feed (calc'd)	100.00	3.42	0.22	100.0	100.0
A-12	2 stage	1000	5	52	3	11.1	Cl conc 2	6.19	50.70	1.10	90.4	25.1
							Cl conc 1	6.89	47.85	1.13	95.0	28.8
							Ro conc	7.73	43.10	1.12	96.0	32.0
							Ro tail	92.27	0.15	0.20	4.0	68.0
							Feed (calc'd)	100.00	3.47	0.27	100.0	100.0

Table 7 (Cont'd)

Test No.	Grind K_{80}	$CuSO_4$ addition g/t	Collector added to rougher g/t		Zn ro skim time, min	Zn ro pH	Zinc flotation results					
			R343	NaAF			Product	Wt %	Analysis %		Distribution %	
									Zn	Pb	Zn	Pb
A-13	2 stage	1000	10	39	3		Cl conc 3	5.55	53.92	0.76	81.6	16.6
							Cl conc 2	6.37	52.94	0.85	92.0	21.4
							Cl conc 1	6.60	51.74	0.89	93.1	23.1
							Ro conc	9.10	39.00	0.90	96.8	32.1
							Ro tail	90.90	0.13	0.19	3.2	67.9
							Feed (calc'd)	100.00	3.67	0.25	100.0	100.0
A-14	2 stage	1000	10	52	3	11.3	Cl conc 3	6.15	52.83	0.78	90.0	18.8
							Cl conc 2	6.59	51.18	0.83	93.4	21.5
							Cl conc 1	6.97	48.81	0.85	94.2	23.2
							Ro conc	10.03	34.89	0.80	96.9	31.4
							Ro tail	89.97	0.13	0.20	3.1	68.6
							Feed (calc'd)	100.00	3.61	0.26	100.0	100.0